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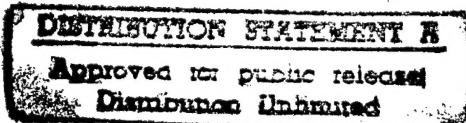
SOVIET NONFERROUS METALLURGY

No. 7

SELECTED TRANSLATIONS

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Introduction

This is a serial publication containing selected translations on nonferrous metallurgy in the Soviet Union. This report contains translations on subjects listed in the table of contents below.

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## 1. Operational Procedure at the Kirovgrad Concentrator Plant\*

*[This is a translation of an article written by S. I. Popov, D. A. Bolkov and L. D. Kulikov in Tsvetnyye Metally (Nonferrous Metals), No. 12, December 1959, pages 1-6; CSO:3001-N/7 (1).]*

The Kirovgrad Concentrator Plant was activated in 1931 and in its first few years its processing of ores involved an incomplete recovery of all valuable components. For instance, the ores of the Karpushinskiy Deposit were processed to obtain copper, zinc and pyrite concentrates, but the quality of these concentrates was low -- as was the degree of recovery of metals from them.

Despite the high content of zinc and sulfur in the ores of the Levikhinskiy Deposit, these ores had been processed so as to obtain the copper concentrate alone. In this connection, the zinc and sulfur used to be irreversibly forfeited in the dump tailings.

It was only in 1942 that a scheme of bulk flotation of all sulfides with secondary grinding of the bulk concentrate and its segregation into copper and pyrite concentrates, in a strongly alkaline medium, was proposed for the ores of the Levikhinskiy Deposit.

At operation according to that scheme (Fig. 1) the ore yielded a copper concentrate containing 12.02-12.86 percent Cu and 12.65 percent Zn, with the copper being recovered to the extent of 88.07 percent. In this connection, the zinc was basically floated together with the copper concentrate.

The consumption of principal reagents at operation according to that scheme was as follows (in kg/ton of ore): xanthogenate (85 percent) -- 0.116; flotation oil -- 0.100; cyano-melt -- 0.123; and lime -- 3.94.

The scheme of bulk-selective flotation proved to be more progressive than the previously applied ones, because it made it possible, upon a more coarse grinding of the ore prior to its bulk flotation, to increase the plant's product-

\*The studies of the plant described in the present article were conducted with the active collaboration of Senior Scientific Research Worker of the Uralmekhanobr /Ural Scientific Research Institute for Mechanical Concentration of Minerals/ G. V. Kozlov.

Upon the introduction of the bulk flotation scheme the researchers at the plant and from the Uralmekhanobr Institute began to work on the separation of the bulk concentrate into copper, zinc and pyrite concentrates. On the basis of the obtained results the Mekhanobr had designed supplementary equipment for the concentrator plant. However, inasmuch as this equipment was not built, the research was reoriented in the direction of recovering zinc from the bulk concentrate into copper-zinc concentrate for the subsequent processing of the latter by the pyroselective method.

The technological scheme and regime of flotation were being worked out on taking into account the equipment possessed by the plant. In May 1955 this scheme was installed and the plant began to provide the copper-zinc product.

The need for the plant's operation according to this scheme was caused by the difficulty in obtaining quality-grade zinc concentrate from zinc-poor ores, and by the lack of experience in operation according to the bulk-selective flotation scheme.

During the period when the plant operated according to the scheme yielding the copper-zinc product, experience had been accumulated and workers had been trained in operation according to a more complicated scheme. At the same time, the technological regime and the scheme itself were being continually perfected. Thus, the year 1956 witnessed the installation and activation of the scheme of triple-quintuple reclari-fication of the zinc concentrate, which made it possible to increase substantially the content of zinc in the monometal concentrate, but the content of copper had remained high because of the transition of a part of the copper from the copper cycle of flotation to the zinc one.

A way out of the resulting difficulty was found by feeding all the bulk concentrate into a ball mill and introducing the alkalization of the pulp to the extent of 200-300 grams of free calcium oxide per cubic meter of the basic copper flotation.

The introduction of this improvement into the bulk flotation practice excludes to some extent a total suppression of sphalerite in the copper cycle, but at the same time it makes possible a satisfactory solving of the problem of the recovery of copper.

Considerable attention was devoted to the problem of testing various schemes for the secondary grinding of bulk concentrates under industrial conditions, to wit: the scheme with the feeding of bulk concentrate to the classification cycle and dispatching of classifier sands to a mill and of classifier overflow -- to copper flotation; the scheme with the dispatching of the bulk concentrate to hydrocyclones with subsequent secondary grinding of hydrocyclone sands and of the

sands of the spiral classifier in a ball mill and dispatching of the overflow of the hydrocyclone and classifier to copper flotation; and the scheme with the dispatching of the bulk concentrate (without thickening) to a ball mill operating in a closed cycle with a spiral classifier, upon dispatching the classifier's overflow to copper flotation.

Of the three tested schemes the last one proved to be most effective, as it made it possible, because of the revivification of the faces of the grains of the entire bulk concentrate, to improve the action of the depressor reagents on the minerals of zinc and pyrite, which was conducive to a better separation of sulfides, zinc and iron.

As a result of the work conducted on improving the scheme of the selective flotation of bulk concentrate (Fig. 2), throughout the entire 1958 the plant supplied only quality-grade zinc concentrate.

The indexes of the processing of impregnated ores according to the bulk-selective flotation scheme and year are presented in Table 1, which shows that in measure with the improvement in scheme and technological regime the qualitative indexes of concentration have improved and the amount of the issued quality-grade zinc concentrate has increased; the amount of the copper-zinc product has decreased till it reached zero in 1958. The net profit gained by the plant as a result of the realization of the zinc concentrate in 1958 totaled several million rubles.

Table 1

Indexes of the Processing of Impregnated Ores According to the Bulk-Selective Flotation Scheme

Year	Content, in percent, of				Content in		Recovery of Zinc	
	Cu		Zn		Copper-Zinc		in percent	
	in Copper		in Zinc		Product, in		Into	Into
	Concentrate		Concentrate		% of		Zinc	Zinc
	in Copper		in Zinc		Cu	Zn	Product	Concen-
	Concentrate		Concentrate					trate
1955	11.55	12.34	--	--	8.07	27.36	44.36	--
1956	11.77	6.93	2.54	48.89	5.16	36.25	31.86	33.07
1957	11.14	8.32	1.59	50.93	3.60	40.68	43.91	42.47
1958	10.47	8.45	0.67	50.86	--	--	-	52.62

1. raw ore (impregnated)	26. sands
2. I stage of grinding	27. basic Zn flotation
3. I classification	28. coarse Zn conc.
4. overflow of rake classifier	29. thickening
5. sands	30. thickening
6. II classification in spiral classifier	31. filtration
7. overflow of spiral classifier	32. I reclarification
8. sands	33. conc.
9. basic bulk flotation	34. tailings
10. grinding	35. overflow
11. coarse bulk concentrate	36. $FeS_2$ conc.
12. dump tailings	37. basic Cu flotation
13. reclarification of bulk conc.	38. finished Cu concentrate (machine No 30)
14. reclarified bulk conc.	39. II reclarification
15. other products	40. conc.
16. coarse Cu flotation	41. III reclarification
17. reclarification of Cu. conc.	42. conc.
18. Conc. Cu flot.	43. IV reclarification
19. grinding in ball mill of O section	44. conc.
20. Cu conc. reclarification	45. V reclarification
21. other products	46. conc.
22. Cu, other products	47. tailings
23. Cu tailings	48. thickening
24. classification in spiral classifier	49. filtration
25. overflow of O-section spiral classifier	50. finished Zn concentrate

The collective of the concentrator plant devotes considerable attention to reducing its operating expenditures. Thus, in 1958 a major project was conducted with regard to reducing the consumption of the principal reagent -- xanthogenate and flotation oil. This made possible not only a partial reduction in the consumption of other reagents but also an improvement in the qualitative indexes of concentration for both the bulk and the selective cycles of flotation.

At present, the plant consumes 42 grams of butyl xanthogenate and 43 grams of flotation oil per ton of processed ore, whereas before the conduct of that project the consumption rates of these reagents used to amount to 88 and 75 grams per ton of ore, respectively. The economic effect yielded by the reduction of the consumption of reagents in 1958 amounted to nearly one million rubles.

The introduction of the scheme of thickening in hydro-

cyclones and the simultaneous broadening of the front of thickening and filtration has raised steeply the yield of pyrite concentrate, which in turn made it possible to convert the Combine's sulfuric acid shops to the processing of a cheaper and qualitatively better raw material and to discontinue the importing of pyrite from other enterprises.

The economic effect yielded by the conduct of these measures in 1958 had expressed itself in the sum of about two million rubles.

From the foregoing it is clear that in the last few years the collective of the Kirovgrad Concentrator Plant has been successfully solving the problems of a comprehensive utilization of the raw material and reduction in operating costs.

However, as a result of imperfections of the scheme of selective flotation, the lack of a sufficient supply of flotation machinery, and for a number of other reasons, the process has often been violated; the copper concentrate still continued to have a high content of zinc and a 10-11 percent content of Cu, and zinc was not recovered from the sulfidic ores containing from one-and-one-half to two percent Zn.

In 1959 the work on the further perfecting of the concentration process was continued -- January witnessed the introduction of the scheme of the joint processing of bulk concentrates obtained from impregnated and sulfidic ores into copper, zinc and pyrite concentrates (Fig. 3).

According to this scheme, the bulk concentrate from impregnated ores is subjected to secondary grinding (to size of 87-90 percent in the -0.074 mm class) in a ball mill and thereupon subjected to selective flotation, during which the following are removed at the beginning of the process: finished copper concentrate ("copper head"), coarse copper concentrate, and middlings. The coarse copper concentrate is subjected to double reclarification together with the copper-zinc concentrate obtained from sulfidic ore.

The concentrate of the control copper flotation is, jointly with the tailings of the first copper reclarification, classified in a hydrocyclone into overflow and sands. The sand fraction proceeds for secondary grinding to a ball mill. The overflow of the hydrocyclone is directed to a thickener so as to remove excess water; thereupon it proceeds to re-flotation; the froth product of the re-flotation is sent to first reclarification of coarse copper concentrate, as this product contains the minerals of copper and zinc. The tailings of the re-flotation, which contain a high amount of zinc are together with coarse zinc concentrate subjected to re-clarifications.

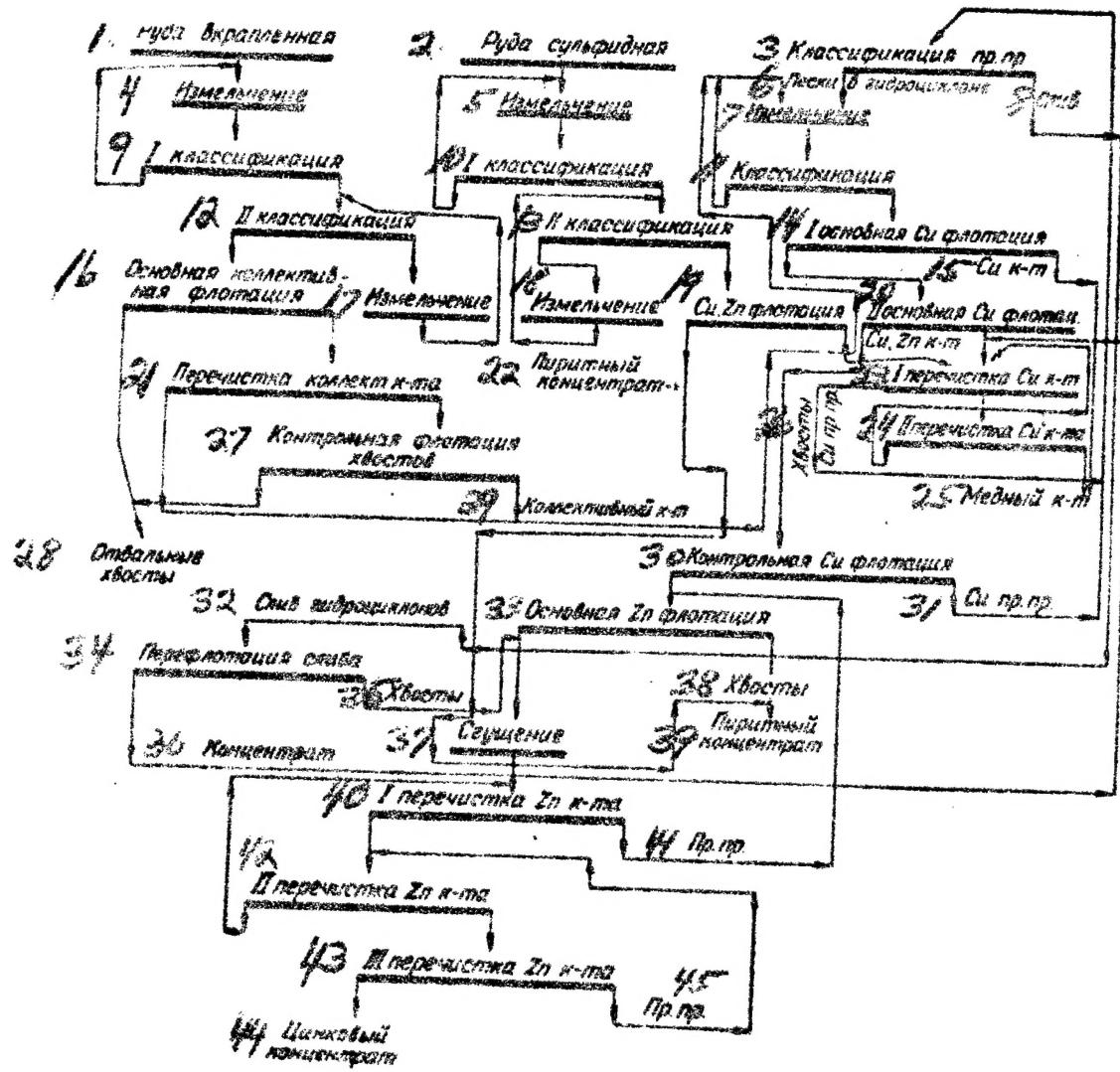


Fig. 3. Schematic Representation of the Processing of Impregnated and Sulfidic Ores, Yielding Copper, Zinc and Pyrite Concentrates

1. impregnated ore	23. I reclarification of Cu conc.
2. sulfidic ore	24. II reclarification of Cu conc.
3. classification of other products	25. Copper conc.
4. grinding	26. tailings Cu, other products
5. grinding	27. control flotation of tailings
6. sands in hydrocyclone	28. dump tailings
7. grinding	29. bulk conc.
8. overflow	30. control Cu flotation
9. I classification	31. Cu, other products
10. I classification	32. overflow of hydrocyclones
11. Classification	33. basic Zn flotation
12. II classification	34. refloating of overflow
13. II classification	35. tailings
14. I basic Cu flotation	36. concentrate
15. cu conc.	37. thickening
16. basic bulk flotation	38. tailings
17. grinding	39. pyrite concentrate
18. grinding	40. I reclarification of Zn conc.
19. Cu, Zn flotation II basic	41. other products
20. Cu flotation	42. II reclarification of Zn conc.
Cu, Zn conc.	
21. reclarification of bulk concentrate	
22. pyrite conc.	
	43. III reclarification of Zn con.
	44. zinc concentrate
	45. other products

The tailings of the copper cycle proceed to basic zinc flotation, the frother product of which is, after its triple reclarification, issued as finished concentrate. The tailings of the basic zinc flotation and the tailings of the copper-zinc flotation from sulfidic ores constitute the pyrite concentrate.

The technological indexes of the processing of the bulk concentrates of impregnated and sulfidic ores according to the selective flotation scheme are cited in Table 2.

Table 2

Indexes of the Processing of Bulk Concentrates  
in 1959 (in percent)

Product	April			May		
	Cu	Zn	S	Cu	Zn	S
Copper Concentrate	14.0	3.58	38.93	14.24	4.65	38.31
Zinc Concentrate	1.03	48.96	34.48	1.14	50.22	35.03
Pyrite Concentrate	0.24	0.62	45.67	0.28	0.67	45.45
Dump Tailings	0.11	0.42	6.69	0.08	0.30	5.73
Recovery of Copper into Mono-Metal Concentrate, in percent:						
from impregnated ore	88.32	--	--	87.92	--	--
from sulfidic ore	80.8	--	--	81.48	--	--
Recovery of zinc into Monometal Concentrate, in percent:						
from impregnated ore	--	55.10	--	--	55.03	--
from sulfidic ore	--	32.02	--	--	33.71	--
Recovery of Sulfur into Pyritic Con- centrate, in percent	--	--	59.85	--	--	61.91

Specific Features of the New Scheme

1. The combined processing of the bulk concentrates of sulfidic and impregnated ores, which makes it possible to organize the recovery of zinc from all the ores handled by the plant.

2. Incorporation into the scheme of the cycle of selective flotation of hydrocyclones for the thickening and sizing of the copper middlings with the subsequent reflootation of the overflow of hydrocyclones; this makes it possible to extract a considerable part of zinc and pyrite with a low content of copper, after the copper cycle of flotation.

The introduction of the above measure has reduced the amount of material sent for basic copper flotation; this has been favorably reflected in the results of the selection of copper and zinc minerals and it has improved the stability of

the recovery of metals into monometal concentrates, which has risen somewhat above the plan-set targets.

3. The organization of the double reclarification of the coarse copper concentrate from impregnated ores and the copper-zinc concentrate from sulfidic ores; this has ensured a two- or three-percent rise in the content of copper in the monometal concentrate as a result of the improved depression of zincous and pyritic minerals.

4. The newly introduced scheme is a higher-developed one, and thus it permits a fuller utilization of the flotation properties of minerals.

5. The practice of the plant's operation according to the new scheme has proved the feasibility of the separation of copper-zinc concentrates obtained from sulfidic ores in a highly alkaline medium into copper and zinc concentrates, upon their joint processing together with the products of the copper cycle of flotation, containing insignificant amounts of alkali (up to 50-100 grams of free CaO per cubic meter).

In the years 1959-1965 the collective of the Kirovgrad plant will continue its work on the further broadening of the comprehensive utilization of processed ores.

It will be necessary to solve the problems of raising recovery of: copper from impregnated ores -- to 91 percent, and from sulfidic ores -- to 85 percent; zinc from impregnated ores -- to 65-70 percent, and from sulfidic ores -- to 40-45 percent; and sulfur into pyrite concentrate -- to 65 percent.

At the same time it is necessary to strive for increasing the copper content of copper concentrate by 1.4 times against the present level.

The problems of the further expansion of the comprehensive utilization of ores will be solved through: development of new schemes and technological regimes; introduction of new technological processes of ore concentration; replacement of old, worn equipment by new, up-to-date equipment; and automation of the process of ore grinding and of the control and regulation of the feeding of reagents.

## 2. The Technology of Concentrating Copper Ores of the Udokansk Deposit

*[This is a translation of an article written by A. I. Izmodenov, S. L. Rokhlin, and G. I. Demidovich in Tsvetnyye Metally (Nonferrous Metals), No. 12, December 1959, pages 19-22; CSO:3001-N/7 (2).]*

On the territory of Chitinskaya Oblast a new large copper deposit was discovered and is now being explored within the confines of the Udokansk Range (Bibl. 1-3).

From the genetic standpoint, the copper mineralization appears to be a sedimentary one. The series of cupriferous sandstones is crumpled into the so-called Udokansk Syncline. The dip of the extended southern and northern limbs of that syncline ranges from 30 to 50 degrees. The northern limb of the deposit is the most significant one.

The deposit displays a notable development of processes of secondary sulfidic enrichment and a zone of oxidized ores.

In their composition of copper minerals the cupriferous sandstones of the Udokansk Deposit are classified into sulfidic, oxidized and mixed. Tests of their concentrability were initiated in 1957 at the Uralmekhanobr /Ural Scientific Research Institute for the Mechanical Concentration of Minerals/.

The investigated three ore samples represent a finely impregnated copper ore whose metallic part consists of the following copper-bearing minerals: chalcosine, covellite, chalcopyrite, bornite, malachite, cuprite. The principal nonmetallic minerals are quartz and, to a smaller degree, feldspar and muscovite.

The extent of the concretions of metallic minerals ranges within the broad limits of from 0.0016 to 2 mm. Concretions measuring 0.01 to 0.09 mm predominate.

According to the results of a chemical analysis (Table 1), the investigated samples contained 1.31-2.78 percent of copper and 65-71 percent of silica.

It can be seen from the rational analyses of the investigated samples that the ores of the Udokansk Deposit differ in their composition of cupriferous minerals: sample No. 1 consisted mainly of secondary copper minerals (98 percent); sample No. 2 was characterized by the presence of secondary (62, 42 percent) and oxidized (25.56 percent) minerals; sulfates are present also; and, as in sample No. 1, a small amount of primary copper sulfides is present.

Table 1  
Results of Chemical Analysis

Element	Percentile Content, %		
	Sample No. 1	Sample No. 2	Sample No. 3
Copper	2.60	1.31	2.78
Zinc	0.02	0.29	Traces
Lead	0.048	0.0057	0.012
Sulfur	0.86	0.59	2.86
Cadmium	--	0.049	0.03
Cobalt	--	Traces	Traces
Molybdenum	--	Not detected	0.001
Iron	3.42	3.03	4.00
Silica	67.75	71.75	65.51
Alumina	12.23	9.72	12.20
Calcium Oxide	1.70	3.28	2.66
Magnesium Oxide	1.01	0.36	0.46
Titanium Dioxide	0.36	0.38	0.54
Manganese Dioxide	0.068	0.10-- 0.13	0.10-- 0.13
Miscellaneous	--	3.40	4.0
Selenium	--	0.0006	0.0006
Tellurium	--	Traces	Traces

Sample No. 3 contains primary (48.9 percent) and secondary (45.52) copper sulfides and, to a smaller degree, oxidized (4.04 percent) and sulfate (1.47 percent) copper.

An assaying analysis showed the following content of gold and silver in the samples:

No. of sample	content, in grams/ton, of	
	Au	Ag
1	0.1	13.4
2	Traces	11.5
3	"	5.41

The conducted tests were oriented toward determining the simplest and most efficient technological schemes of copper recovery.

The presence of secondary sulfidic and oxidized cupriferous minerals inclined to form slimes and the polydisperse character of their dissemination predicate the necessity of applying concentration schemes that involve multiple-stage grinding and interstage flotation.

The following principal schemes were tested:

- (1) Scheme of single-stage flotation;
- (2) Scheme of two-stage flotation (Fig. 1);
- (3) Scheme of three-stage flotation (Fig. 2);
- (4) Scheme of two-stage flotation with return of intermediate product to the second stage of flotation and second stage of grinding.

Laboratory investigations established that the two-stage flotation illustrated in Fig. 1 is an efficient scheme of concentrating the ores of the Udkansk Deposit. Upon coarse grinding of ore to 57-62 percent in the -0.074 mm size and flotation in an alkali lime medium (pH = 8.9-9.1) with sodium sulfide and butyl xanthogenate, it is possible to obtain during the first stage of flotation a finished high-grade copper concentrate containing 35 percent Cu, with the recovery of copper amounting to 62.2 percent for sample No. 2 and 82 percent for sample No. 3.

Thus the lion's share of copper is already extracted at coarse grinding during the first stage of flotation, and the resulting copper concentrate is of good quality. After the secondary grinding of the tailings of the first stage of flotation to 96 percent in the -0.074 mm size, it is possible to recover an additional 28.1 percent Cu into copper concentrate containing 11 percent (sample No. 2) and 13.68 percent Cu into copper concentrate containing 13 percent (sample No. 3).

The ultimate technological indexes obtained for the tested ore samples are cited in Table 2.

The fine grinding of ore (to 96 percent in the -0.074 mm class) is an indispensable prerequisite for obtaining quality grade copper concentrate.

The composition of the copper concentrate includes the following principal components: copper -- up to 22 percent; silica -- 33-39 percent; alumina -- seven-eight percent; iron -- six-12 percent; calcium oxide -- three percent; and silver -- 20-70 grams/ton. The recovery of silver into the copper concentrate reaches 57-59 percent.

The mineralogical and chemical analyses of classes of the dispersion analysis of flotation tailings showed that copper is lost both in the fine classes (-10 microns), because of overgrinding, and in the coarser class of 43 microns, because of undergrinding. The copper lost in the

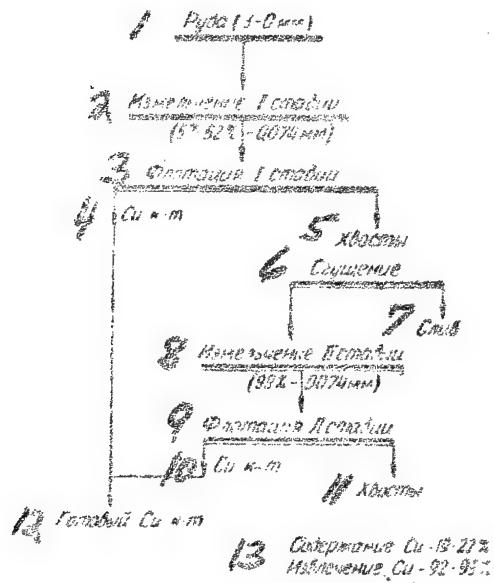


Fig. 1. Schematic Representation of the Two-Stage Flotation of the Copper Ores of the Udekansk Deposit

1. ore (3-0 mm)	7. overflow
2. first-stage grinding (57.62% -0.074 mm)	8. second-stage grinding (99% -0.074 mm)
3. first-stage flotation	9. second-stage flotation
4. Cu conc.	10. Cu conc.
5. tailings	11. tailings
6. thickening	12. finished Cu conc.
13. Content of Cu: 19-22%	
Recovery of Cu: 92-95%	

tailings is mostly in the form of oxidized compounds (sample No. 2 -- as much as 75 percent of all the copper thus lost; and sample No. 3 -- up to 46 percent). In sample No. 3 a major part of the lost copper is lost in the form of secondary (30 percent) and primary (23 percent) copper sulfides.

The total consumption of reagents for each of the tested schemes is cited in Table 3.

The schemes of two- and three-stage flotation were followed while conducting experiments with the utilization of the reflux liquor in the process of grinding and flotation. This led to establishing the possibility of obtaining

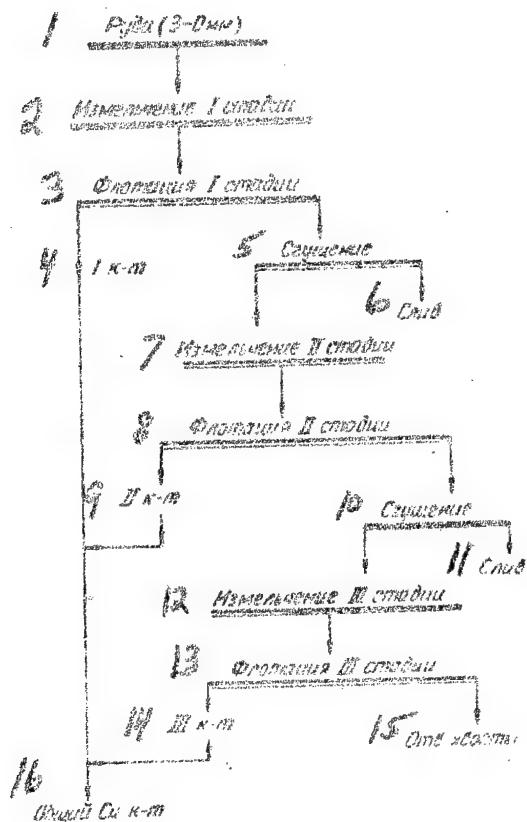


Fig. 2. Schematic Representation of Three-Stage Grinding and Flotation

1. ore (3-0 mm)	9. second conc.
2. first-stage grinding	10. thickening
3. first-stage flotation	11. overflow
4. first conc.	12. third-stage grinding
5. thickening	13. third-stage flotation
6. overflow	14. third conc.
7. second-stage grinding	15. dump tailings
8. second-stage flotation	16. total Cu conc.

identical results of flotation and simultaneous reducing the consumption of reagents.



Table 3

Total Consumption of Reagents by Tested Scheme,  
in grams/ton

Reagent	Two-Stage Flotation	Three-Stage Flotation
Sodium Sulfide	600-900	1,500
Lime	1,600-1,300	1,500
Buty Xanthogenate	80	110
Pine Oil	40	60

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### 3. The Roasting of Pelletized Copper Concentrates and Charges in a Fluidized Bed at a Semi-Industrial Installation\*

This is a translation of an article written by E. Ya. Serebrennikova in Tsvetnyye Metally (Nonferrous Metals), No. 12, December 1959, pages 23-28; CSO:3001-N/7(3). ]

In 1953 the Gintsvetmet /State Institute of Non-ferrous Metals/ had conducted studies of the fluidized-bed roasting of copper concentrates and charges in an enlarged continuous-action laboratory furnace. Subsequently, this research was continued in a semi-industrial installation with a furnace hearth measuring three  $m^2$  in area, constructed at the SUMZ /Sredneural'sk Copper Smelting Plants/. The results then obtained served as a foundation for designing and constructing a roasting furnace at that plant.

Inasmuch as the fluidized-bed roasting of fine-disperse copper concentrates has its disadvantages in the form of a considerable carry-over of dust in the furnace and a comparatively low furnace productivity (seven or eight tons/ $m^2$  daily), therefore in 1955 the Gintsvetmet had conducted studies of the process of the pelletizing of copper concentrate and the fluidized-bed roasting of the pelletized copper concentrate.

The conducted laboratory experiments and subsequent semi-industrial tests demonstrated the broad prospects existing for the pelletizing of charge prior to its roasting. The first experiments with fluidized-bed roasting at the experimental installation of the SUMZ were conducted with pelletized copper concentrate obtained by drying in tubular kilns.

\*Study conducted with the consultation of Cand. Eng. Sci. G. Ya. Leyzerovich; the investigations of the pelletizing of charges were conducted by Gintsvetmet researchers Cand. Eng. Sci. A. A. Orionov and Engineer V. S. Pyzhov in collaboration with the workers of the experimental shop of the SUMZ  
\*\*G. Ya. Leyzerovich, I. V. Babina, and E. Ya. Serebrennikova, "Tsvetnyye Metally," No. 6, 1955

The concentrate fractions used in the experiments were sifted on two screens to sizes of from 0.5 to four mm.

The development of the regime of roasting of the pelletized concentrate was accompanied by tests of the designs of heat-transfer devices applied at operation with high unit loads.\*

#### Flow Sheet of the Pelletizing Roasting Installations

By the end of 1957 the construction of a semi-industrial pelletizing installation was completed. The charge, consisting of concentrate, quartz and limestone, with a moisture content of 8-10 percent, was conveyed from the plant's drying shop to the bin of that installation and thence carried by a tray feeder to a breaker consisting of two cast toothed drums rotating in opposite directions. The teeth of one drum mesh with the teeth of the other so as to leave a clearance of two to three millimeters, so that the material falling in between the teeth is broken up into small pieces which proceed to a turret-bowl pelletizer with a diameter of 1-1.5 meters and productivity of 1,200-1,300 kg of dry pellets per hour; 80 percent of the pellets thus obtained measure from one-half to three mm in size.

The moisture necessary for binding the pellets was supplied to the pelletizer by means of a sprayer. The moist pellets proceeded to a drying kiln with a hearth measuring  $0.5 \text{ m}^2$  in area.

The drying of the pellets was conducted in a fluid bed by flue gases obtained from the combustion of solar oil and mazut.

The dust-laden gases emitted by the drying kiln were, after cleaning in two cyclones, expelled by a fan into the atmosphere.

The unit productivity of the drying kiln amounted to 70-80 tons of charge per square meter daily, and its rate of moisture abstraction amounted to approximately  $300 \text{ kg/m}^2 \text{ per hour}$ . The temperature in the bed was  $70^\circ\text{C}$ ; the temperature of the flue gases prior to their entry into the drying kiln amounted to  $300-315^\circ\text{C}$ . The residual moisture content of the pellets after drying was 0-1 percent. The consumption of gases for drying amounted to 1.65 standard cubic meters per kilogram of charge.

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\*I. A. Burovoy, G. Ya. Krichevskiy, and A. D. Shtokarev, "Tsvetnyye Metally", No. 3, 1958

The dry pellets are conveyed by two transporters to the bin of a roasting furnace from which they are subsequently conveyed by a feed belt to a weight doser and thence dropped through a chute into the furnace. The furnace itself, after its dimensions were reduced, had a chamber measuring 1x0.7 meters and an overflow-threshold height of 1.2 meters. The bottom of the furnace was built of heat-resistant concrete. The gases proceeded through a vertical pipe with a diameter of 808 mm. Their purification was initially conducted in two cyclones with a 0.8-mm diameter, and later in the Cottrell filter of the drying shop. After purification the gases were expelled into the atmosphere and the roasted residue or calcine was, after weighing in hopper cars, conveyed to a depot. The cyclone and flue dusts were weighed and sent to the charge department of pelletization for use as a binding agent.

The air from the turbo blower was conveyed to air-distributing boxes and thence through nozzles into the furnace (Fig. 1). The open area of the orifices in the nozzles is one percent. The resistance of the furnace sole without the fluid bed at the necessary amount of air ( $1,300 \text{ m}^3/\text{hour}$ ) amounts to 200 mm of the water column.

Heat transfer was realized by spiral pipes positioned vertically or horizontally inside the fluid bed at a height of 180-200 mm above the furnace sole (Fig. 2). Their specific area amounted to  $3.7 \text{ m}^2 \text{ per } \text{m}^2$  of furnace sole, at a 1,200-mm bed height. Despite the considerable saturation of the bath by heat-absorbing elements the process of the pseudo-liquefaction of the material was not hindered and the technological regime was stable.

Six thermocouples were installed to measure temperature in the layer. One of these thermocouples was interlocked with an automatic emergency injector of water into the bed actuated in the event of a rise in temperature to  $850^\circ\text{C}$  and disconnection of the pumps feeding water to the spiral pipes.

#### The Effect of Cyclone Dust on the Strength of Pellets at Roasting in Fluidized Bed

To obtain strong charge pellets it is necessary to introduce the following admixtures in the process of pelletizing.

Considering that the fluidized-bed roasting of pelletized material involves the formation of a disperse dust containing up to 10 percent of the aqueously soluble salts, the possibility of utilizing that dust as a binding agent during the pelletizing of charge was investigated.

The strength of pellets was tested during the

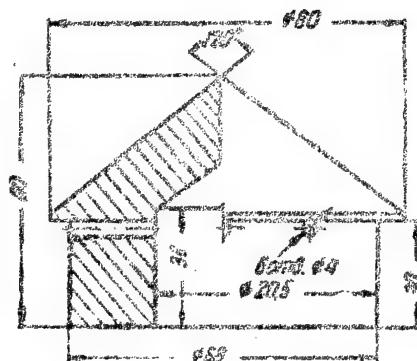


Fig. 1. Air-Distributing Nozzle

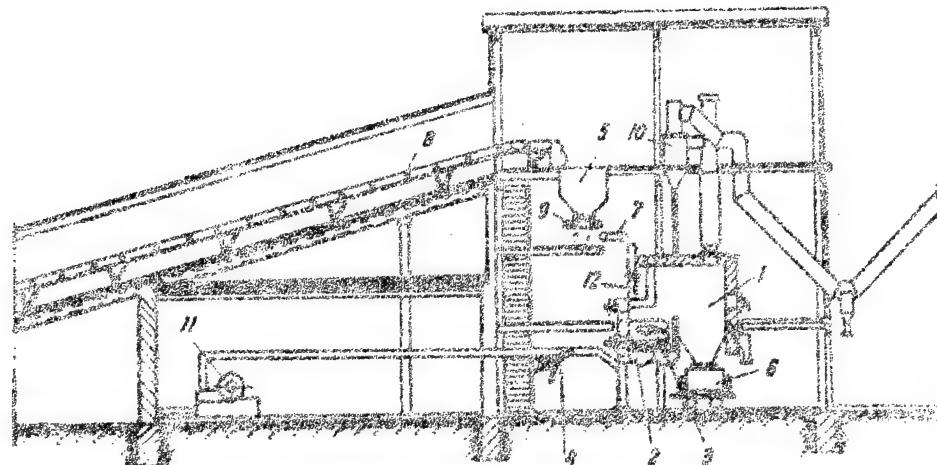


Fig. 2. General View of the Experimental Roasting Installation

1. Furnace	7. Weight doser
2. Furnace sole	8. Transporter
3. Heat-abstracting devices	9. Feed belt
4. Burner for the start-up of the furnace	10. Cyclone
5. Charge bin	11. Turbocompressor
6. Hopper car	12. Chute

fluidized-bed roasting in an enlarged continuous-action laboratory furnace constituted by a stack of heat-resistant steel measuring 106 mm in diameter and 3,500 mm in height. An inclined chute for the overflow calcine was installed on the 1,000 mm level.

Air was fed in through an air-distributing nozzle. The charging was conducted by means of a sluice feeder. The dust-laden gases were purified in a cyclone and a bag filter.

The experiments were conducted with a pelletized charge consisting of concentrate, quartz, limestone, and cyclone dust, the last being taken in the proportion 5, 10 and 15 percent to the weight of the charge (Table 1). The temperature during the experiments was 800°C, the linear air-feeding rate was 47.5 cm/second, and the unit productivity of the furnace amounted to 38.5-42.5 tons/m<sup>2</sup> daily. The SO<sub>2</sub> content of the gases amounted to 13-14 percent, and the sulfur content of the calcine amounted to 4.5-7.5 percent. The carry-over of dust from the furnace amounted to 11-15 percent of the weight of its burden. The strength of the pellets after roasting was satisfactory, and their mean weighted diameter changed from 0.99 to 0.76 mm.

Table 1

Granulometric Composition of Charge, in percent

Material	Screen-Hole Size, in mm					Mean-Weighted Diameter of Grains, in mm
	-2/2	/1.2	/0.7	/0.5	-0.5	
Charge	11.02	12.7	46.8	20.5	9.0	0.99
Calcine	7.33	8.12	39.25	22.8	22.5	0.76

Thus, laboratory studies had established the feasibility of using the cyclone dust obtained from the fluidized-bed roasting of pellets as a binding agent for pelletization.

Heating and Start-Up of the Furnace

The solving of the problem of raising the productivity of the furnace required a considerable enlargement of the surface of the device for abstracting excess heat, which complicated the heating of the charge bed during the

start-up of the furnace. This circumstance had necessitated the development of a special regime of the heating and start-up of the furnace. This regime consisted in the following.

A solar-oil burner was mounted in the air conduit along which the air from the blower enters the furnace. The air for spraying solar oil was fed by a compressor with a pressure of 1.2-2 atmospheres (Fig. 3).

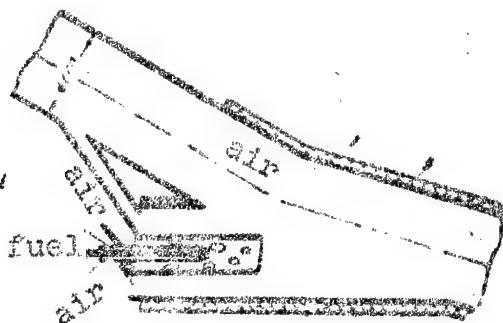


Fig. 3. Installation of Burner for Activating the Furnace

1. Air duct into the furnace
2. Burner for heating the air entering the furnace
3. Peephole
4. Tube with orifices for directing the flame
5. Lining of air duct

The burner heated the air entering into the process to a temperature of 250-300°C, whereupon calcine was placed in the furnace.

After the temperature of the bed had reached 400-450°C the burner was disconnected and the charging of raw burden was begun. Within 40-45 minutes after the activation commenced the bed had acquired the set temperature of 800°C.

The start-up of the furnace was simple and took little time in contrast with the start-up of furnaces involving the heating of the bed by burners mounted behind the fluidized bed, such as is practiced in industrial furnaces.

Such a start-up can be recommended for the industrial furnaces too.

### Characteristics of the Raw Material

The experiments were conducted with a pelletized concentrate. Pelletized charge consisting of concentrate, limestone, and crushed gold ore. The binding agent used was sulfite-pulp alkali in the proportion of 0.8 and 1.2 percent to the weight of the charge, or cyclone dust (15 percent of the weight of the charge). Later on, all experiments were conducted with a charge pelletized together with cyclone dust.

Tables 2 and 3 cite the chemical and granulometric compositions of the charge.

Table 2  
Chemical Composition of Charge

Product	Percentile Content of				
	Cu	Zn	S <sub>Common</sub>	S <sub>SO<sub>4</sub></sub>	SiO <sub>2</sub>
Charge Pelletized with Alkali	8.4	9.7	32.7	0.33	18.3
Charge Pelletized with Cyclone Dust	8.15	9.2	30.2	0.77	20.8
Concentrate	10.1	10.6	40.6	0.3	4.0

Table 3  
Granulometric Composition of Charge

Screen-Hole Size, in mm	/3	/2	/1	/0.4	-0.4	Mean-Weighted Diameter of Grains, mm
Yield of Fractions, in mm	4.35	38.7	35.15	10.6	11.2	1.74

## Experimental Part

The experimental furnace at the SUMZ was used to work out the regime of the roasting of pelletized material, to study the behavior of pellets during roasting in fluidized bed, and to test out new designs of heat-absorbing devices with automatic regulation of the process.\*

The technological indexes obtained during the fluidized-bed roasting of pelletized charge are compared with the analogous indexes for unpelletized charge in Table 4.

From Table 4 it can be seen that the unit productivity at roasting of pelletized charge is four and one-half times as high as at roasting of unpelletized charge, and the carry-over of dust one-third to one-fourth as low.

During roasting, the coefficient of the transfer of heat from the bed to the wall of the heat exchanger installed in the fluid bed ranged from 150 to 180 kilocalories/m<sup>2</sup> hour degree, which is less than the heat transfer coefficient in the bed of unpelletized fines (200-220 kilocalories/m<sup>2</sup> hour degree). Such a decline in the heat transfer coefficient in the former case is related to the decrease in the density of the fluid bath, as a result of the increase in grain sizes and the considerably higher velocity of the flow of air necessary for converting large grains to a pseudo-liquid state.

The post-roasting residue (calcine) contained as much as 5-15 percent S; 9.5-10.5 percent Cu; 10-11.8 percent Zn; and 18-20 percent SiO<sub>2</sub>. The cyclone and flue dusts contained 8-11 percent S; 8-10 percent Cu; 10-11 percent Zn; and 18-20 percent SiO<sub>2</sub>.

The dusts were sent to the pelletizing department for blending with raw charge.

Although the calcine was basically represented by pellets of a sufficient strength, a part of the pellets had nevertheless previously, during roasting, disintegrated as a result of the decrepitation of the pyrite included in the composition of the charge, and as a result of attrition in the fluid bed. The dust yield amounted to 15-20 percent of the burden, i.e. to the amount necessary for pelletizing.

The calcine obtained from the roasting of pelletized charge had a granulometric composition that differed from that of the calcine obtained from the roasting of unpelletized charge (Table 5); thus, the >0.4 mm fraction of the pelletized calcine contained 12 percent of grains with sizes

\*Conducted in collaboration with the workers of the experimental shop, SUMZ

Table 4

Comparative Technological Indexes of the  
Roasting of Pelletized and Unpelletized Charge

Index	Charge	
	Pelletized	Unpelletized
Roasting Temperature, in 0°C	800	800
Air Consumption Rate, in m <sup>3</sup> /kg	1.16	0.95
Cold-Air Linear Feed Rate, in cm/second	48	8.0
Unit Productivity of Furnace in Terms of Charge, in tons/m <sup>3</sup> daily	≤ 38	8.0
Unit Productivity of Furnace in Terms of the Burning-Out of Sulfur, in tons/m <sup>2</sup> daily	to 12	1.8-2
Mean Weighted Content of Sulfur in Calcine, in percent	5-15	9.3
Content of SO <sub>2</sub> in Roasting Gases (from Furnace) in percent of abstraction	14	14
Yield of Calcine Through Overflow Threshold, in percent of Solid Products of Roasting	58.2-62.6	12-20
Carry-Over of Dust in percent of Solid Products of Roasting	19.8-15.4	68-60
Size of the Calcine Discharged Through the Overflow Threshold (mean weighted diameter) in mm	1.14	0.21
Desulfurization, in percent	81-65	70.4

smaller than 0.07 mm, while the same fraction in the unpelletized calcine contained 50 percent of grains with sizes smaller than 0.07 mm.

If, however, it is considered that the yield of calcine from the roasting of unpelletized material amounts to 15-20 percent, and cyclone dust is the principal product of such roasting -- with 90 percent of that dust being of a size smaller than 0.074 mm -- then the difference in the mesh size of calcines will be even greater. It should also

Table 5

Granulometric Composition of Calcines, in percent

Calcine	Screen Mesh in mm					Mean Weighted Diameter in mm
	/3	/2	/1	/0.4	-0.4	
From Pellet- ized Charge	1.46	11.1	32.0	37.0	18.2	1.14
From Unpellet- ized Charge	--	--	1.74	3.36	94.9	0.21

be noted that the pelletizing of a charge consisting of a concentrate and fluxes ensures a proper intermixing of components and their close mutual contact, which is reflected positively in the results of the roasting and during the smelting of the calcine as well.

On the other hand, the roasting of unpelletized charge results in the segregation of fluxes and concentrate (Table 6): the fluxes, being larger in size, remain in the bath and are discharged through the overflow threshold, while a major part of the concentrate escapes from the furnace in the form of dust; in this connection, the dusts thus obtained are rich in copper and zinc and poor in silica (12 percent  $\text{SiO}_2$ ), whereas the calcines are rich in silica (32 percent  $\text{SiO}_2$ ). During the roasting of pelletized charge, the calcine and the dust both contain approximately equal amounts of all metals.

It should be noted that the cyclone dust obtained during the roasting of pelletized charge is larger in size (content of fractions smaller than 0.074 mm: 60-70 percent) than the dust from the roasting of unpelletized charge (content of fraction smaller than 0.074 mm: 90-95 percent).

The expediency of employing pelletization for preparing the charge prior to roasting was corroborated by studies of the performance of an industrial furnace for the fluidized-bed roasting of unpelletized cupriferous charge at a copper-smelting plant in the People's Republic of Bulgaria.

Although the copper concentrate in Bulgaria is ground more coarsely (fractions smaller than 0.074 mm: about 50 percent) than the SUMZ's concentrate (fractions smaller than 0.074 mm: about 95 percent), the process of the roasting of

Table 6

Chemical Composition of Pelletized and Unpelletized Materials, in percent

Nature of Material	Charge			Calcine			Dust		
	Cu	Zn	SiO <sub>2</sub>	Cu	Zn	SiO <sub>2</sub>	Cu	Zn	SiO <sub>2</sub>
Pellet- ized	8.4	9.8	20.2	10.0	10.7	20.2	8.3	10.7	19.6
Unpellet- ized	9.0	6.3	10.1	7.5	6.5	32	9.5	8.0	12

Bulgarian concentrate involves the formation of a considerable amount of fine-disperse dust which leaves the furnace and settles in dust-collecting devices. The collection of all that dust into a single pile and its smelting, and especially its charging into the smelting furnace, cause a number of difficulties.

Upon the roasting of a pelletized charge, on the other hand, the pelletized calcine will be subjected to smelting while the dust will be returned for pelletizing.

The Gintsvetmet has conducted on a laboratory scale studies of the pelletizing, roasting and smelting of copper charge at the Alaverdi Copper-Chemical Combine, which yielded positive results.

The investigations conducted in 1958-1959 on the semi-industrial installation made it possible to work out all the technological parameters of the processes of pelletizing and of the drying and fluidized-bed roasting of pellets, and to provide the Unipromed [Scientific Research Institute of the Copper Industry] with data for an operative designing of the roasting shop at SUMZ.

- END -

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